LIXIVIATION IN-SITU DE L'OR

G. Deschênes
Hydrométallurgiste

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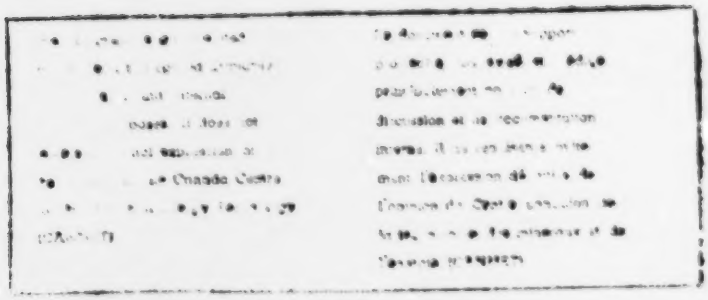

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LIXIVIATION IN-SITU DE L'OR

par

G. Deschênes*

RÉSUMÉ

La lixiviation de l'or est un sujet de plus en plus populaire. Des travaux australiens ont permis de faire progresser ce concept. Quelques commentaires sur le projet australien sont présentés dans ce rapport. De plus l'auteur discute brièvement la partie de la revue relative à la lixiviation in-situ de l'or faite par Chamberlin.

Mots-clés: Or, in-situ, lixiviation, cyanure, thiourée

Keywords: Gold, in-situ, leaching, cyanide, thiourea

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*Hydrométallurgiste, Laboratoire de métallurgie extractive, Laboratoires des Sciences Minérales, CANMET, Energie, Mines et Ressources Canada.

INTRODUCTION

Un projet sur la lixiviation in-situ de l'or a été présenté par P.R. Whincup dans le cadre, "World Gold '89". Ce forum sur l'or a eu lieu à Réno, du 5 au 8 novembre dernier.

P.R. Whincup fait partie d'une companie, nommée BHP Utah Minerals International, dont le siège social est situé à Melbourne en Australie. Cette compagnie a initiée un programme d'exploration, en 1981, de systèmes de paléodrainage ternaire localisés en Australie australe.

Les résultats d'exploration ont permis de caractériser une zone "palaeochannel". Les canaux sont localisés de 60 à 100 m sous la surface du sol et se présentent sous forme de lits d'anciennes vallées de rivières. Ces canaux sont formés de gravelle, de sable, d'argile et d'eau souterraine à forte salinité.

Les caractéristique du gisement et la faible teneur en ore (0.03 à ~ 2 g/t) ne permettent pas de procéder au minage conventionnel du dépot. C'est ainsi que BHP Utah Minerals Int. a décidé de démarrer une étude de lixiviation in-situ car cette compagnie considérait que c'était le seul moyen d'exploiter cette propriété.

Différents lixiviants ont été évalués pour récupérer l'or du "placer". Le chlore, le brome, le thiosulfate, la thiourée et le cyanure forment ce groupe. Différents raisons techniques ont amené BHP Utah Minerals International à retenir le cyanure.

La figure 1 nous montre un schéma du système de lixiviation in-situ. Les canaux de circulation de la solution ont été aménagés dans la zone visée et servent à acheminer le lixiviant sous-terre et à le retirer. Il a été constaté qu'il est techniquement possible de récupérer l'or à cause de facteurs favorables. Parmi ces facteurs, nous avons: (1) le faible écoulement de l'eau souterraine; (2) une zone imperméable existant sous le gisement et permet dès lors de récupérer la plupart de la solution injectée; (3) l'or est accessible à la solution de lixivation; (4) les sédiments d'or sont perméables; (5) la solution ne cause pas des dommage irréversibles à l'environnement aquatique; (6) la faible teneur en composantes cyanicides des sédiments.

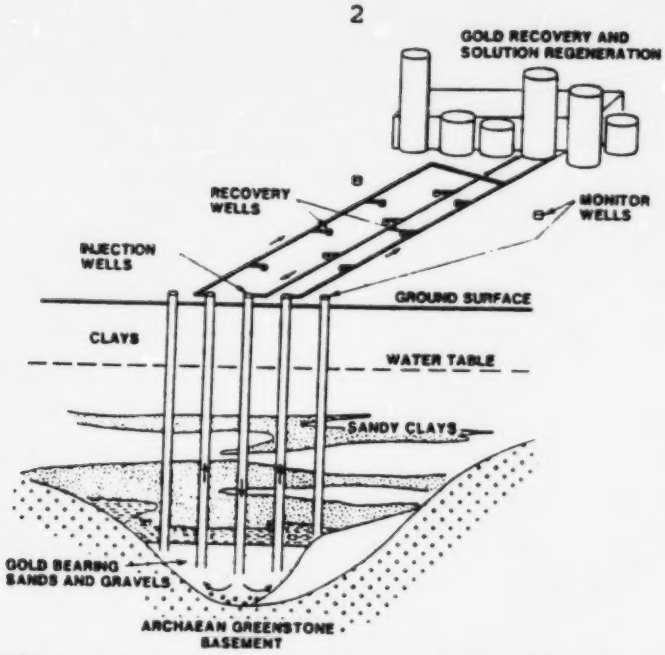

Fig. 1. Diagramme de traitement de l'or par lixiviation in-situ

La forte salinité de l'eau souterraine a cependant été la cause de l'échec de ce projet. En effet, la présence de Mg^{2+} provoque la précipitation de brucite ($Mg(OH)_2$) lorsque la concentration en cyanure de la solution excède 200 ppm. De plus, la solubilité de l'oxygène est fortement réduite dans une telle solution.

Par conséquent, le système de lixiviation possèdait une cinétique très faible. Il s'est avéré que la vitesse d'extraction de l'or est un facteur important dans la rentabilité du projet. Ainsi la lenteur de récupération a rendu le projet inacceptable pour atteindre l'échelle industrielle.

Un copie de la publication est annexée à ce rapport et donne de plus amples détails sur les travaux réalisés en Australie.

Une revue sur la lixiviation par percolation (heap, dump and in-situ) a été faite par P.D. Chamberlin. Une copie de ce rapport est également annexé.

Dans le cas qui nous concerne, l'auteur discute des différents projets de lixiviation in-situ. Parmi les différents métaux et composés traités, nous retrouvons l'uranium, le cuivre, le sel, la potasse et le soufre. Un mélange de thiosulfate d'ammonium et de carbonate d'ammonium a

été utilisé par Jacobson et Murphy pour traiter un gisement d'argentite. Un composé d'iode est utilisé pour traiter les dépots d'or "placer". L'iode est regénéré par électrolyse avec une cellule à diaphragme tout en récupérant l'or.

Chamberlin croit que la lixiviation in-situ de l'or a le potentiel pour atteindre le stage de commercialisation mais que le dommage environnementale causé par le cyanure ralentira le projet. De plus, il déclare que cette méthode de lixiviation percera lorsque les autres lixiviants de l'or seront commercialement acceptés.

CONCLUSIONS

Les travaux australiens ont prouvés qu'il est techniquement possible de récupérer l'or d'un gisement "palaeochannel" par une technique de lixiviation in-situ utilisant le cyanure. Le système n'a pas eu d'impact nocif sur l'environnement. Toutefois le projet n'atteindra pas l'échelle industrielle à cause de la lenteur de procédé. Il aurait été intéressant d'évaluer plus à fond la thiourée car elle possède une meilleure cinétique de lixiviation que le cyanure.

L'utilisation de cyanure, quoi qu'étant favorisée par les conditions du projet australien, ne serait pas souhaitable pour la lixiviation souterraine dans une mine; la principle obstacle étant les émanations éventuelles de HCN qui compromettent sérieusement la sécurité du personnel.

Les travaux australiens ne sont pas directement applicables aux conditions des gisement filonniens mais certains éléments de leur approche vont être surement très utiles dans notre projet.

Chapter 29

In Situ Leaching of Western Australian Palaeochannel Gold

Whincup, P.R.[1], McDonald, D.C.[1], Duke, H.[2]

[1]BHP-Utah Minerals International, Asia Pacific Division, Melbourne, Australia
[2]BHP Engineering, Sydney, Australia

ABSTRACT

Due to grade, overburden depth and stability and the presence of groundwater it was considered impractical to recover gold contained in unconsolidated palaeochannel sediments by conventional mining, dredging or borehole mining techniques.

A series of field experiments was conducted to determine the feasibility of recovery of gold by in situ leaching from sediments located at a depth of about 80 metres.

The test wellfield site and plant are described together with field test procedures, results and wellfield rehabilitation procedures. Choice of sodium cyanide as leachant and laboratory testing are discussed in relation to groundwater and sediment chemistry and problems encountered in field tests.

INTRODUCTION

Placer gold deposits within Tertiary palaeochannel systems of the Yilgarn Block in Western Australia collectively represent a mineral resource which although low in gold concentration could provide the basis for a substantial new gold mining venture in that State. The channels, described in detail by Smyth and Button, (1989), reach between 60 and 100 metres below the surface and represent old river valleys which have filled with unconsolidated gravel, sand, clay and groundwater of high salinity.

Due to depth, stability of overburden and the presence of groundwater at a shallow depth, techniques including open cut mining, borehole mining and dredging would be both impractical and uneconomic. Desktop studies were conducted which showed that in situ leaching offered the only possibility for technical and commercial success in exploitation of this resource.

In situ leaching has been applied for a number of years to extraction of uranium and copper and although there have been proposals for in situ leaching of gold ores dating back to 1896 [Mineev and Shutov, (1979)], little work to develop the concept is known.

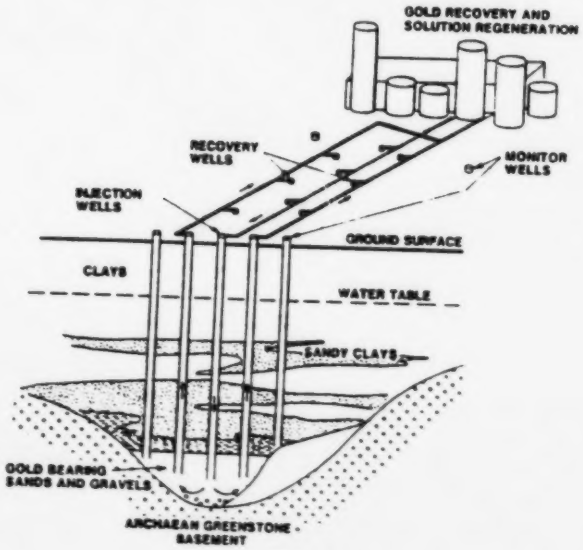

Figure 1
Conceptual Diagram for In Situ Leach Operation

Figure 1 shows a conceptual in situ leaching system for extracting gold from auriferous alluvial deposits. The commonly postulated requirements of such a deposit for application of in situ leaching are:

- The size of the deposit and its recoverable gold content are such that treatment is economic.
- For all practical purposes the aquifer containing the gold bearing sediments should be static or slow flowing.
- Ideally, impermeable zones should exist above and below the gold bearing sediments.
- Gold particles must be accessible to the leach solution.

- The leach solution should not irreversibly damage the aquifer environment.

- The sediments should be low in minerals other than gold which consume or otherwise adversely affect the leach reagent.

- The gold bearing sediments should be permeable.

In 1985 BHP-Utah Minerals International commenced a program of laboratory and field testwork to determine the feasibility of in situ leaching of gold from Western Australian palaeochannels involving auriferous sediments of the Rebecca palaeochannel.

LEACH SYSTEMS

Development of a leach system suitable for in situ leaching of Tertiary Channel sediments was a process involving consideration of channel groundwater quality, initial laboratory lixiviant screening testwork and the solution of problems which arose during field testwork.

Groundwater Quality

Typical analysis of groundwater contained in the auriferous sands and gravels is shown in Table 1. The water is saturated with respect to sodium chloride, gypsum and brucite [$Mg(OH)_2$].

Table 1

Typical Analysis of Groundwater Associated with Gold Bearing Sediments in the Rebecca Palaeochannel

Component	g/l
Na^+	83
K^+	1.12
Ca^{2+}	0.61
Mg^{2+}	8.5
Cl^-	143
SO_4^{2-}	16.6
Total Dissolved Solids	307
pH	6
Density	1.165g/cm^2
Temperature	22°C (on site - September 1985)

The buffering capacity of the magnesium prevents the pH rising above 8.7 unless lime is added in excess of 35 kg/tonne to precipitate magnesium ions.

With groundwater of this quality the choice of leachant chemistry must be such that significant increase in calcium, sulfate or hydroxyl ions does not occur. It was believed raising either the calcium or sulfate concentration would lead to formation of gypsum, which would probably coat particle surfaces, thus limiting lixiviant access to gold particles. Similarly an increase in pH was shown to restrict solution flow through sediments as a result of precipition of gelatinous brucite.

Leach Reagent Selection

Laboratory testwork was conducted to choose a suitable lixiviant for in situ leaching. Solutions of the lixiviants cyanide, bromine, chlorine, thiourea and thiosulfate in test site groundwater were trialled using gold strip. Consideration was also given to the interaction of each reagent with other minerals occurring in the sediments, the groundwater and its environmental acceptability.

Bromine and chlorine were excluded because oxidation of sulfides present in the sediments would increase sulfate concentration and produce gypsum which had been observed to coat gold particles.

Thiourea required acid to reduce groundwater pH and reagent costs would be high. The carcinogenic nature of thiourea also provided grounds for rejection of this reagent.

Thiosulfate showed low gold dissolution rates in the laboratory tests and a reasonably high concentration of reagent would be required to stabilise the gold complex in solution. Breakdown of thiosulfate could also lead to gypsum formation.

The foregoing considerations resulted in the cyanide/oxygen system being chosen as the preferred lixiviant. In addition, the chemistry of gold cyanide complex formation is relatively well understood; breakdown of cyanide was unlikely to irreversibly damage the aquifer environment; and, if necessary, cyanide retained in the aquifer could be destroyed by mild chlorination.

However, there were problems associated with the choice of cyanide/oxygen as the lixiviant including:

- Toxicity of cyanide.
- An elevated pH is usually necessary to maintain cyanide in solution.
- Low solubility of oxygen in the high salinity groundwater.
- The tendency of cyanides to form hydrogen cyanide (HCN) in high salinity groundwater with subsequent loss of HCN to atmosphere.

When sodium cyanide was added to the groundwater the pH was found to increase to 8.7 at which point brucite precipitated, buffering the solution. Laboratory experiments showed that a maximum of 200ppm sodium cyanide could be dissolved in the groundwater before brucite precipitation commenced and therefore this level was suggested as optimum for field tests from the point of view of avoiding the possibility of pore

and wellscreen blockage.

Nevertheless, concentrations of sodium cyanide higher than 200ppm were used in the initial field tests as it was assumed fine diffuse precipitates would not block pores. However at the highest cyanide concentration used, 1,500ppm (0.15% w/v) sodium cyanide, nucleation and growth of brucite quickly plugged the well screens demonstrating the reality of the anticipated problem.

In groundwater of the composition shown in Table 1, solubility of pure oxygen was found to be about 3ppm maximum at 1 atmosphere which is only about 30% of that of well aerated carbon-in-leach circuits. Thus the diffusion rate of oxygen to gold particles would be slower in the groundwater than for aerated water of low total dissolved solids. Consequently some field tests incorporating oxygen injection were undertaken.

The high ionic strength of the groundwater and pH buffering both contributed to enhanced chemical stability of HCN and high partial pressure of HCN caused cyanide losses on exposure of solutions to air. It was suggested that any cyanide system using high salinity groundwater would have to be totally enclosed to limit cyanide loss.

Use of calcium cyanide or pH adjustment with lime was considered inappropriate as these would promote formation of gypsum.

Laboratory column leach tests using groundwater and auriferous sands taken from exploration bores in the Rebecca palaeochannel were undertaken in 100mm NB x 1 metre high PVC columns. Various concentration levels were used in these tests. An additional column, packed with clean quartz sand with a gold plate of known surface area, was used as a control and to assess gold leach rates.

From this testwork a possible extraction of about 12% of the gold from channel sands was estimated for a 14 day contact period using a solution containing 200ppm sodium cyanide solution with air as oxidant. The rate at which gold leached using the 200ppm sodium cyanide solution was estimated to be 0.3 mg/day/cm^2.

TEST SITE

A field test site was established over a section of the Rebecca palaeochannel on sparsely vegetated arid pastoral land located some 135 kilometres east north east of the gold mining centre of Kalgoorlie, Western Australia.

Gold had been located within sands and gravels immediately above impermeable Archaean saprolitic greenstone bedrock at a depth of about 80 metres. The channel contained groundwater, of composition shown in Table 1, with the water table approximately 7 metres below ground level.

WELLFIELD CONSTRUCTION

A wellfield comprising 6 wells designed for small scale push-pull and circulation testing was established in a pattern designed for testwork using flows over varying distances, both along and across the channel.

Wells were constructed using conventional water well techniques. The wells were rotary mud drilled to a point approximately 5 metres below the unconformity at the base of the Tertiary alluvial sequence. Each hole was then cleared of drilling debris and gamma logged. The gamma logs provided accurate identification of the unconformity and were used to locate the position for placing well screens. Each hole was reamed, fitted with 155 mm NB Class 9 PVC casing and a 1 metre long x 155 mm NB x 0.5 mm aperture, wedgewire screen located immediately above the unconformity.

Test wells incorporated a neoprene gasket immediately above the screen and gravel packing which allowed the annulus above the screen to be sealed with cement to prevent direct leakage between the target horizon adjacent to the screen and those horizons higher in the sequence.

HYDROLOGICAL CONSIDERATIONS

The objective of hydrological testing was to determine the necessary parameters to be able to model the extent of lixiviant flow within the sediments.

From regional flow studies of the Rebecca palaeochannel area, intrinsic groundwater velocity was established as about 1 metre per year. A very low groundwater velocity was confirmed by tracer test and leaching test results. This very low velocity reduced the complexity of modelling and monitoring of field testwork.

Field tests were performed to allow an assessment to be made of the hydraulic conductivity of the gold bearing sediment by three different methods; flooding bores at constant and falling heads; by using drawdown data and a two dimensional computer model; and by use of a three dimensional model developed by In-Situ Inc., of Laramie, Wyoming. Values determined varied significantly with method used. However, modelling results provided by In-Situ Inc. were considered to present the most accurate representation of the aquifer under test. A mean hydraulic conductivity of 4.1 m/day over a section of thickness 15.1 m was accepted. Table 2 shows that the value determined is highly dependent on the model chosen to represent the groundwater flow regime.

The nature of deposition of cross bedded river channels in general leads to anisotropy of permeability and hydraulic conductivity parameters. The possibility of channelling of reagents through the coarse sediments (for example in the thalweg of the river bed) was cause for concern about the variations in hydraulic

conductivity within the sediments. Such variations would facilitate leachant excursions and hinder solution access to the finer grained material.

Table 2

Hydraulic Conductivites of the Aquifer at the Test Site

	Base Depth m	Thickness m	Hydraulic Conductivity m/day	
Australian Groundwater Consultants	80.5	0.6	47.8*	22.3**
	81.4	0.6	0.5*	0.2**
	82.3	1.0	4.0*	2.0**
	83.5	1.0	7.2*	8.1**
BHP Engineering 2 Dimensional Model	82.0	2.0	20.0 Average+	
In-Situ Inc. 3 Dimensional Model	82.0	15.1	4.1 Average+	

* falling head tests
** constant head tests
\+ determined as best fit of drawdown and pump test data

FIELD TESTWORK

Field testwork undertaken comprised push-pull tests conducted with both high and low strength lixiviant, two-well circulation tests and an extended four-well circulation test. Push-pull and two-well circulation tests using a lithium tracer were also conducted.

Push-Pull Tests

Push-pull tests were undertaken to establish the methodology for testwork and to determine if recovery of gold from palaeochannel sediments using in situ cyanide leaching was possible.

Performance of push-pull tests and interpretation of results are relatively simple as these tests involve small volumes under static conditions and use of a single well. For these reasons push-pull tests were chosen as the starting point for field testwork.

Push-pull tests comprised injection of a volume of about 25 m^3 of leach solution into the aquifer and allowing it to remain for several days before withdrawal of liquor from the same well. Samples of recovered liquor were analysed to determine gold recovery, lixiviant consumption and identify chemical reactions which took place between leach solution and aquifer sediments.

The circuit shown in Figure 2 was used for push-pull tests.

Prior to commencement of a push-pull test groundwater was pumped from the wellfield into

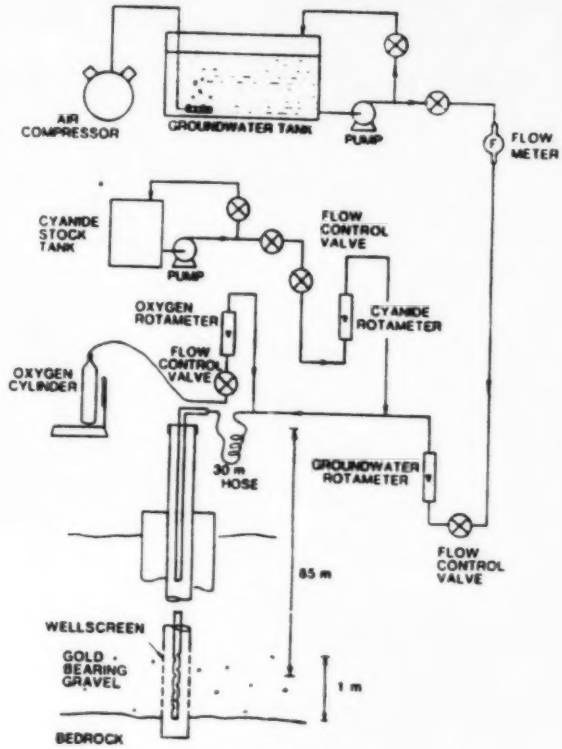

Figure 2
Equipment Configuration for Push-Pull Testwork

one or more holding tanks, each of about 40 m^3 capacity. A nominal 5% w/v sodium cyanide stock solution was mixed in a separate tank using fresh water.

Leach solution was introduced into the well at a rate of approximately 40 l/min. using a 25 mm diameter plastic pipe positioned at the well screen. Cyanide stock solution was added to groundwater stock immediately prior to introduction into the well.

Rotameters, flow control valves and in-line flow meters were used to monitor and adjust flow rates but measurements of volumes of liquor used were made by measurement of liquid levels in respective tanks.

For early tests only aerated groundwater was used in the leach solutions. However, in later tests gaseous oxygen was introduced into the leach solution immediately prior to injection into the well. In some tests a quantity of oxygenated groundwater was introduced into the aquifer ahead of the leach solution as a buffer to maintain a high aquifer oxygen level in the lixiviant pulse.

Solution was generally allowed to remain in the aquifer for about 6-7 days before recovery. In some tests the leach solution was moved or

'pulsed' daily by drawing out about 2 m^3 and replacing it with fresh lixiviant or by simply injecting an additional 2 m^3 of fresh lixiviant.

Solution was recovered by lowering a well pump to a point approximately 40 m above the well screen and pumping at the same rate at which the solution had been introduced, that is nominally 40 l/min. Samples were taken at 15 minute intervals for the first two hours of recovery and half hourly thereafter. Each sample was immediately subject to cyanide, Eh, pH and oxygen activity determination and a portion set aside for gold, iron and copper analysis.

Oxygen activity was determined using a TPS PTI-401 dissolved oxygen meter. Owing to the high ionic strength of the solution, determinations made in this way could not be considered a quantitative measure of the dissolved oxygen but simply a qualitative measure of relative oxygen activity.

Recovered liquor was stored in surface tanks for later disposal into a surface evaporation pond after passing through activated carbon columns for gold recovery.

Tracer Test

At the commencement of the program a push-pull test was conducted using lithium chloride tracer. The purpose of this test was to demonstrate that, in conducting a push-pull leach test, a high recovery of injected liquor could be achieved. Lithium chloride was chosen as the tracer as lithium was not intrinsic to the groundwater. Also, lithium ions did not react appreciably with the sediments and accurate analysis at low lithium concentrations could be performed on site using atomic absorption spectrophotometry.

Other tracers such as tritium and bromide were considered but rejected due to the extended time and extra equipment required for appropriate analysis of these ions.

An accurately measured volume of solution containing a known quantity of lithium chloride was introduced into the test bore at a nominal rate of 40 l/min. over a 9 hour period.

The injected liquor was allowed to remain underground for 7 days and recovered at a nominal rate of 40 l/min. over a period of 37 hours during which time periodic measurements of volume recovered were made and samples for lithium analysis taken.

The recovery of the lithium tracer as a function of volume of liquor extracted is shown in Figure 3. Approximately 83% of the tracer was recovered in the first pore volume extracted with a total of 94% in two pore volumes and 96.8% after almost four pore volumes. These results demonstrated that for a push-pull test a high recovery of liquor injected could be achieved.

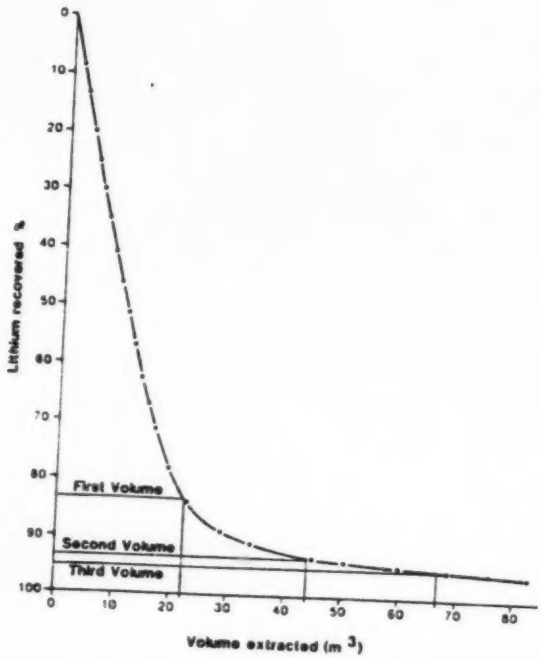

Figure 3
Lithium Tracer Remaining Underground During Recovery Pumping in a Push-Pull Test

Results

A total of 16 push-pull cyanide leach tests was conducted. Initial tests were conducted at a nominal lixiviant strength of 0.15% w/v sodium cyanide. However, following problems associated with well screen plugging attributed to brucite formation in these push-pull tests and in early two-well circulation tests, all further testwork was undertaken using nominal sodium cyanide solution concentrations in the range of 0.01-0.02% w/v.

Test conditions and results of four of the more significant push-pull tests are summarised in Table 3.

Figure 4 shows gold concentration in recovered liquor as a function of volume recovered. Sustained concentration of gold occurred in only one case.

Results from these tests showed:

- Recovery of gold from palaeochannel sediments using in situ cyanide leaching was possible.
- Oxygen activity played a significant role in the dissolution process.

Table 3

Summary of Conditions and Results of Cyanide In Situ Leach Push-Pull Tests

Test No.	B5	B6	B9	D21
Volume injected - m^3	22.2	24.6	29.0	25.0
Rate of injection - l/min.	41.1	36.4	44.1	31.4
Lixiviant strength - % w/v sodium cyanide	0.135	0.133	0.155	0.021
Oxygen added to lixiviant	No	Yes	Yes	Yes
Oxygen buffer	No	No	Yes	Yes
Residence time - hrs	166	166	120	226
Pulsed	No	No	No	No
Volume recovered - m^3	86.1	37.9	66.9	58.7
Recovery rate - l/min.	37.4	43.4	40.5	37.4
Average sodium cyanide concentration of first pore volume recovered % w/v sodium cyanide	0.062	0.061	0.073	0.008
Relative oxygen activity of first pore volume recovered	5	2	21	5
Average pH of first pore volume recovered	8.59	8.67	8.89	8.21
Calculated gold recovered - mg	723	913	2610	1355
Calculated iron recovered - g	181.2	113.3	46.8	26.9
Maximum solution gold tenor - ppm	0.03	0.06	0.08	0.08
Calculated cyanide recovery - %	59.3	49.6	54.3	51.1

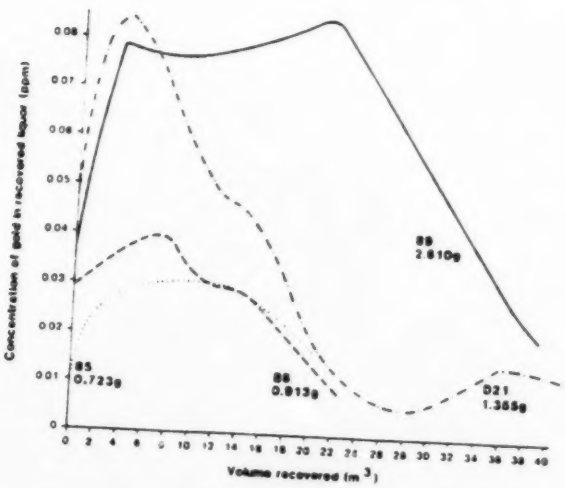

Figure 4
Gold Recovery for Selected Push-Pull Tests as a Function of Solution Volume Recovered

- At least 50% of injected cyanide was recovered. (It is believed significant HCN loss occurred from the high salinity, oxygenated groundwater during the sample taking process or during oxygenation.)
- A 0.02% w/v sodium cyanide solution will dissolve gold from the sediments but, as expected, the dissolution rate appears lower than that for a 0.15% solution.

Two-Well Circulation Tests

Following several push-pull tests conducted at a lixiviant strength of 0.15% w/v sodium cyanide, a program of two-well circulation leach testing was undertaken to establish the methodology for two-well circulation testing and to demonstrate that lixiviant flow between wells could be established.

The circuit shown in Figure 5 was set up and operated intermittently over a period of 14 days. Numerous delays to operation in the circuit were caused by mechanical failure of equipment. Periods of restoration pumping aimed at limiting excursion of lixiviant from the injection well were also undertaken. In all, the circuit was operated in the circulation mode for some 82 hours with an additional 75 hours total of restoration pumping.

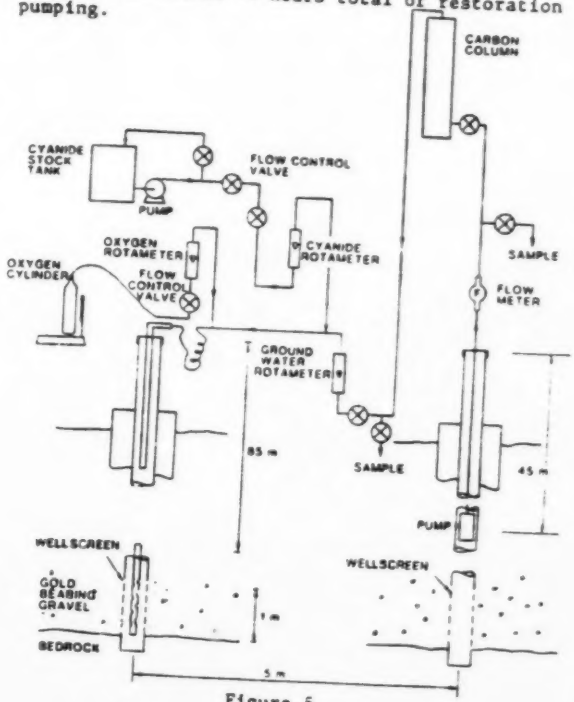

Figure 5
Equipment Configuration for Circulation Leach Tests

During operation of the circuit, significant well screen scaling and plugging of carbon columns occurred due to brucite formation. This resulted in a steadily increasing circuit pressure and decreasing solution flow.

Tracer Test

In order to establish that a high recovery of injected liquor was possible during circulation and restoration pumping phases of a two-well circulation test, a test using a lithium chloride tracer was undertaken. In this test an accurately measured weight of lithium chloride was dissolved in a known quantity of groundwater and introduced into the injection well at 35 l/min.

Extraction of liquor took place at a well 5 metres distant at a rate of about 50 l/min. and extracted liquor was pumped directly to the waste water disposal area. Circulation was undertaken over 3 pump/rest cycles, each cycle comprising 12 hours pumping and 36 hours rest. Following circulation, extraction continued until the level of lithium in extracted liquor fell below the limit of detection. Results showed that 93% of injected lithium was recovered (see Figure 3).

Results

Test conditions and results for two-well circulation testing are summarised in Table 4.

Table 4

Summary of Two-Well Circulation Test Conditions and Results

Total volume circulated - m^3	153.7
Average injection rate - l/min.	31.2
Injected lixiviant strength- % w/v sodium cyanide	0.15
Oxygen addition rate - g/m^3	132
Circulation time - hrs	82
Elapsed time - hrs	336
Volume extracted in restoration pumping - m^3	212
Calculated gold recovered during circulation - mg	4,415
Calculated gold recovered during restoration pumping - mg	8,381
Calculated cyanide recovery - %	31.3
Relative oxygen activity of extracted liquor	25
Maximum solution gold tenor - ppm	0.13

This work established the ability to pump lixiviant between two wells while maintaining oxidising conditions and confirmed the ability of a cyanide solution to recover gold from the sediments.

During circulation testwork using lixiviant containing 0.15% w/v sodium cyanide, significant operational problems were encountered due to brucite formation. This resulted in carbon column, well screen and possible sediment plugging leading to difficulty in maintaining solution flow in the circuit. This led to the conclusion that an in situ leaching operation in water of quality found in the test site using cyanide concentrations of the order of 0.15% w/v sodium cyanide would not be possible.

Four-Well Circulation Test

The final test undertaken in the program was a four-well circulation test, the primary purpose of which was to establish the methodology for future multiple well tests.

A circuit comprising one extraction and three injection wells was operated for a period of 30 days on daily cycles of 12 hours pumping and 12 hours rest. During the period 590 m^3 was circulated while maintaining an injected lixiviant strength of 0.01% w/v sodium cyanide. During circulation a calculated 9.479 g of gold was recovered.

Following completion of this test, restoration of the total test area was undertaken during which a further 12.570 g of gold was recovered.

Overall recovery of cyanide for the four-well circulation test was calculated to be 50.4%.

REHABILITATION OF THE WELLFIELD

On completion of leaching tests, wells likely to contain residual cyanide were pumped for over one month.

The extracted groundwater, totalling 893 m^3 was regularly tested for cyanide. When cyanide analysis values reduced to a consistently low level, the wellfield was allowed a rest period of 1.5 months to detect any equilibration of cyanide species between the aquifer material and the groundwater.

Following the rest period, withdrawal of groundwater (3,213 m^3) was continued for over one month, and free cyanide analysis by a modified colorimetric technique was used in determining the period of extraction required.

Finally the groundwater was sampled five times over a day's extraction, in the presence of an independent organisation's representative. A similar five samples were obtained from a well with no history of cyanide contamination. All samples taken were preserved with sodium hydroxide in clear plastic bottles, chilled and forwarded to three independent analytical laboratories for free cyanide, total cyanide and potassium analysis (the potassium concentration was used to maintain a check on sample dilution).

All values for free cyanide and total cyanide were well below 0.5 ppm as sodium cyanide. Comparative cyanide analysis figures are provided in Table 5.

All facilities were dismantled and removed from site and only the capped wells have been left. The area was left to regenerate naturally and on recent site visits regrowth of small ephemeral plants was observed. Most of the desert vegetation present in the area should re-establish over several years.

Table 5

Cyanide Analysis of Restored Wellfield Water and Virgin Wellfield Water

	Free NaCN (ppm)		Total NaCN (ppm)	
	Used Wells	Virgin Wells	Used Wells	Virgin Wells
BHP CENTRAL RESEARCH LABORATORIES				
High	0.33	0.22	0.25	0.12
Average	0.28	0.10	0.19	0.09
Low	0.17	0.05	0.14	0.05
THE CHEMISTRY CENTRE OF WESTERN AUSTRALIA				
High	<0.10	<0.10	0.30	<0.10
Average	<0.10	<0.10	0.30	<0.10
Low	<0.10	<0.10	0.30	<0.10
AMDEL				
High	0.42	0.17	0.20	0.40
Average	0.15	0.08	0.13	0.30
Low	0.07	0.04	0.09	0.12

DISCUSSION

The successful extraction of gold from Tertiary channel auriferous sediments by cyanide in situ leaching was encouraging.

Push-pull tests indicated the presence of recoverable gold in the sediments of the wellfield aquifer. However, it is believed the time the lixiviant was resident in the aquifer was too short for a significant solution tenor to develop. Attempts to increase the liquor tenor were made by 'pulsing' in which lixiviant was moved short distances through the aquifer and, while the flow of solution is laminar, some mixing and transfer of reagents was achieved at a higher rate than by diffusion alone. However, no significant effect was detected. Liquor tenor and total gold recovered were improved by increasing both the quantity of available oxidant and the cyanide concentration.

Push-pull tests with lithium tracer demonstrated that recovery of liquor was possible and thus established the ability to recover injected cyanide solution. Tracer tests also confirmed no significant groundwater flow was present in the channel. The high recovery of tracer indicated cyanide not recovered had been consumed by decompositon, by loss to atmosphere on recovery or by adsorption on the sediment surfaces.

While decomposition or atmospheric cyanide losses would be unlikely to constitute an environmental hazard, adsorption or storage of cyanide in the aquifer could become an environmental issue. For this reason stage restoration pumping with lengthy rest periods between stages was undertaken to determine if equilibration between cyanide species and sediments had occurred. It was, however, found that no significant recharge of groundwater with cyanide species occurred during the rest periods indicating such equilibration had not taken place.

Wellfield restoration, while time consuming, proved clean up of the aquifer was possible. Analysis values for free cyanide remaining in the test wellfield after clean up were found to be sufficiently close to those of virgin groundwater indicating restored groundwater to be essentially free of cyanide species. Had this water originally been potable, it could again be used for animal or human consumption. This standard of groundwater restoration was pursued and achieved to demonstrate that it is possible for an aquifer having potential as a potable water source to be restored after gold recovery by cyanide in situ leaching.

To measure low cyanide values in the highly saline groundwater with confidence, it was necessary to develop suitable cyanide analysis techniques. The standard silver nitrate/potassium iodide titration was adopted for high cyanide concentrations. At levels of cyanide below 20ppm use of rhodanine indicator was found to be preferable.

For very low cyanide concentrations a modified pyridine/barbituric acid colorimetric analysis developed by Doolan, (1989), at BHP Central Research Laboratories was used. After appropriate sample preparation the method was used to detect down to 0.01ppm free cyanide. A standard distillation was used to determine total cyanide.

The values in Table 5 show background measurements (for the virgin groundwater) as high as 0.4ppm sodium cyanide, demonstrating the problems associated with cyanide measurement below 0.5ppm. The groundwater salinity forced a tenfold sample dilution during analysis reducing the sensitivity of cyanide determinations.

Table 5 also shows free cyanide values higher than total cyanide were measured, indicating some aberration yet to be rectified in the determination of free cyanide in saline groundwater.

Murphy, McGrew and Jacobson, (1984), suggested that recovery of gold for a given sediment thickness may be determined by the following equation:

$$\text{Recovered Gold} = [1 - \exp(-k \times \text{time})] \times \text{Accessible Gold Grade} \times \text{Area Accessed}$$

where k is the extraction rate constant for the particular sediments and prevailing conditions.

For push-pull tests, values of k in the range of 0.001 to 0.003 per day were determined while for circulation testing a value of 0.003 per day was determined. These values were of the same order of magnitude as that determined in laboratory column tests (k = 0.009 per day) indicating

reasonable agreement between laboratory and field data.

Calculations using a k value of 0.003 per day and the indicated grade and thickness of Rebecca palaeochannel sediments, together with the order of magnitude estimates of capital and operating costs, showed in situ leaching of this deposit would be uneconomic. These calculations also showed that for the process to be economic a rate constant or leachable gold grade greater by an order of magnitude would be required.

Since the testwork showed the rate constant to be a function of leach solution cyanide strength, attention is now being given to deposits in water of lower salinity and in particular lower in magnesium content.

CONCLUSIONS

Field testwork has shown that technically it is possible to extract gold from W.A. Tertiary palaeochannel sediments by in situ leaching using a cyanide/oxygen lixiviant. The apparent rate of gold dissolution was, as expected, dependent on the quantity of available oxidant, gold, and solution cyanide concentration. Improvement in the leach rate by increasing cyanide concentration is limited by formation of brucite in the highly saline groundwater.

An economically viable gold in situ leaching operation would be dependent on attaining a much higher leach rate than that achieved during the field tests. The possibility of improvement in leach rate rests in either increasing the leach rate constant by using a higher cyanide strength, different lixiviant, or by discovery of higher grade deposits.

In addition to the leach rate, the cost of locating the deposit and the cost and number of wells which need to be established are of critical importance in establishing the viability of any gold in situ leaching operation.

Restoration of the aquifer environment tested with respect to cyanide contamination is possible by pumping for extended periods.

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Chapter 26

STATUS OF HEAP, DUMP, AND IN SITU LEACHING OF GOLD AND SILVER

Paul D. Chamberlin

Chamberlin & Associates
Littleton, Colorado

ABSTRACT

The evolution of heap and dump leaching has followed the development of agglomeration, materials of construction for leach pads and ponds, solution distribution systems, testing techniques, and state regulations. They are now mature processes. Possible future directions and developments are discussed.

In situ leaching of precious metals is waiting to be commercialized. Regulators are reluctant to allow the use of cyanide because of the potential for excursions and operators are reluctant because of the potential for lawsuits. Developments in alternative lixiviants are needed before in situ leaching is accepted.

INTRODUCTION

Heap and dump leaching of gold and silver ores have matured over the past 10 years. They are no longer emerging technologies. Enough has been written and discussed about them to give prudent operators a good chance of success on the first effort. This was not the case 10 years ago when the failure rates were very high (Kappes, 1979).

Heap and dump leaching operations can be quickly put into operation. They have always had this reputation. It is common to have an operation online within a year of AFE approval and even four months has been reported. The environmental setting plays a large role in the schedule and it has become obvious to the industry that early and continual contact with the regulatory agencies is essential to a successful project.

Heap and dump leaching also have a reputation as low cost operations. As a result, low grade ores can be treated. The lowest grade, stand-alone heap leach operation that has been reported is AMAX Gold's Wind Mountain project which has an average ore grade of 0.023 oz gold/tonne. Other low grade operations are Coeur d'Alene's Rochester project which processes an ore containing 0.012 oz gold/tonne plus 1.65 oz silver/tonne and Pegasus Florida Canyon operation which processes 0.026 oz gold/tonne. A profitable low grade operation will have one or more of the following characteristics: >5000 tonnes/day throughput, >80-85% recovery of values, no requirement for crushing run-of-mine ore, and stripping ratios that are <1:1. Some dump and heap leach operations treat only the low grade portion of an ore body. In these cases, their operating costs are often only incremental costs because another process that is used to treat the high grade ore is carrying the bulk of the costs. In these cases, material as low as 0.01 oz gold/tonne has been treat profitably.

In situ leaching, on the other hand, doesn't have a single commercial experience to its credit, yet. This technology was widely used in the uranium industry and acceptance is gaining in the copper industry but it has not found acceptance in the precious metals business.

This paper discusses how heap and dump leaching technology has matured and discusses some of the reasons that projects have failed in the past. It then suggests future directions that heap and dump leaching may take. The paper also takes a look at current thinking for the in situ leaching of precious metals and gives prospects for the future.

HEAP AND DUMP LEACHING - PROGRESS TO DATE

The heap and dump leaching industry has made much progress in its understanding of what causes project failures. Some of the reasons for past failures are discussed in the following paragraphs along with improvements that have made heap and dump leaching the successes they are.

Agglomeration

The merits of agglomeration were not widely recognized until the US Bureau of Mines published their work (Heinen, et al., 1979). Since the application of agglomeration technology, the failure rate of new heap leach projects has decreased dramatically. Agglomeration has not only provided great technological advantage but it has also given heap leaching an air of respectability (Tibbals, 1987). Prior to the Bureau of Mines' work, some operators in the copper vat leaching business, such as Yerington and Inspiration, practiced agglomeration by wetting finely crushed ore before loading it into vats.

This simple wetting gave better permeability than if dry ore was loaded into the vats. The precious metals industry and the USBM built on these early copper experiences.

Agglomeration does not imply that binders must be used. In the precious metals business, it simply means that deleterious fines are bonded to coarser particles long enough to prevent segregation of coarse and fine particles as the heap is being constructed. In some cases where the amount of fines is minimal, this "agglomeration" can be achieved by simply wetting the ore with water. The surface tension of the water is enough to bind fines to coarser particles.

In most cases, however, a binder is needed, particularly if there are many fines, which is loosely defined as being >10%-100 mesh. The standard binder is Portland Type II cement which has good sulfate resistance. All cements hydrate down to temperatures as low as 15°F, although the rate slows considerably as the temperature drops. For air temperatures below 40°F, the use of Type I or Type III cement is helpful because their higher tricalcium aluminate content makes them more reactive. Type III Portland cement develops early high strength which is a benefit for agglomeration in cold weather. However, it has 25% to 50% more surface area per unit of weight than Types I and II and will have a greater tendency to agglomerate onto itself than coarser cements do. Therefore, it must be well blended into the ore before moisture is added to get full benefit of all the cement. Otherwise the cement may ball up and agglomerate on itself and be wasted.

Other binders such as lime, fly ash, or flue dust have been used because they are less costly than Portland cement. But, they are generally not as uniform as cement, either chemically or physically, and they have not performed quite as well. Relatively few operators use these other binders.

Empirical test procedures have been developed to quantify the amount of binder and moisture needed for agglomeration (McClelland, 1988). The results of these tests require careful interpretation. If used blindly, they could result in excessive use of binder.

Several machines for agglomerating have been tried but the most commonly used are drum agglomerators, pug mills, and conveyor belts (Chamberlin, 1986). Generally, the pug mills are best on ores with many fines; conveyor belts are best on ores with few fines; and drums are in between. There are some notable exceptions to this observation, however, such as Cominco's Buckhorn operation.

Heap Leach Testing Procedures

Testing ores for their amenability to heap or dump leaching sounds deceivingly simple, but many former operators can testify to the pitfalls of not performing a rigorous test program. Lists of things to do, and not do, have been compiled over the years and reported (Chamberlin, 1981, 1988; McClelland, 1988). A quarterly newsletter called *Heap & Dump Leaching* is published to inform the industry about techniques and procedures. And countless articles have been published about all aspects of the business. Prudent operators today have a wealth of information that can be tapped to avoid pitfalls.

In the past, it was common to get metallurgical data on recovery of precious metals and then go build an operation. That simplistic approach is no longer applicable because environmental concerns require much more. Process development work is done not only on the recovery of gold and silver from ore, but also on the detoxification of solutions and solids and the attenuation of trace elements that may leak from the circuit. Procedures have been established for attenuation testing (Rouse, 1988) and these are becoming more important as state regulators ask, "what if?" questions.

Leach Pad Construction

Ten year ago, the primary leach pad construction materials were natural soils and clays, sometimes mixed with bentonite, soil cements, or sodium salts to reduce permeability of the compacted mass. More often than not, these pads leaked badly because it was difficult to maintain uniform quality during construction. The labor and equipment necessary to do a good job of construction made the pads quite expensive, also. Then geomembranes came on the scene: PVC, Hypalon, and the present favorite, HDPE. State regulators started implementing requirements that were more strict, such as leak detection systems and redundancy in the leach pad itself. The use of synthetics blossomed into geogrids, geotextiles, and perforated pipe. Leach pads, which started as one-liner structures more than a decade ago, became two-liner structures and now, on rare occasions, even three. Sometimes the leach pad is a composite of liner materials, some synthetic and some natural. The complexity has raised costs from <$4.64/m^2 to more than $23.23/m^2 and designing the leach pads is a major part of any project (Strachan and van Zyl, 1988; Richins, 1988).

There are two basic types of leach pads, permanent and reusable. The permanent pad has a heap built on it which will remain forever. Permanent leach pads are much more common because most ores take months if not years to leach completely. The reusable leach pad has a pile of ore leached on it for a finite period of time and then it is removed and replaced by another pile of fresh ore. Only ores that leach within days or a couple weeks are leached on reusable pads. The reusable pads are generally a multiple layered structure incorporating natural materials, synthetics, and asphalts.

Geotechnical Data

The design of leach pads, heaps, and the solution collection system all require an assessment of geotechnical characteristics of materials. These characteristics include permeability of the ore and construction materials, settlement of the ore and construction materials, and maximum compressive loading on agglomerated ore. The slope stability of heaps has become very important, requiring an understanding of the friction angles of the mater-

ials of leach pad construction, and the ore itself. Years ago, the industry didn't recognize the importance of such data, but today it is commonplace and extensively used in designing facilities (van Zyl, 1987; Strachan and Dorey, 1988).

Water Balance

The industry has been embarrassed many times by operators who failed to adequately account for surface water control, precipitation, and snowmelt when designing their facilities. The resultant spills or unintended discharges have brought retaliation by state regulators, and the entire industry suffers.

The knowledge to design adequate leach pads, heaps, and ponds is available (Hutchison, 1988), but the biggest problem is the adequacy of the meteorological data base that is available (or not available). Some operators feel they have been subjected to 100-year storms back to back. They probably haven't, but when pond levels are rising and mud has shut down the mining operations, it is natural to wish that Mother Nature would comply with the historical records used in the process design and be more kind and gentle.

Heap Building

A major reason for heap and dump failure has been poor heap building techniques. Improper heap building methods can create areas of compacted ore, cause extreme segregation of particle sizes, and cause lower and slower gold-silvery recovery (Chamberlin, 1981; Muhtadi, 1988). For example, (1) driving trucks all over the surface of a heap is sure to compact the ore, and (2) building cones of unagglomerated ore with a stacker is sure to create segregation. Many methods of heap building have been tried. Stackers are gaining popularity and are a good way to build heaps provided they move continuously and lay down narrow windrows of ore. Truck and dozer systems are also common but they require careful control of truck traffic and deep ripping between lifts.

Solution Distribution

Leach solution has been applied to the surface of heaps and dumps by several devices over the years. The two most favored devices today are Wobbler sprinklers and drip irrigation systems (Dixon, 1988; Chamberlin, 1988; Muhtadi, 1988).

Typically the leach solution is percolated through the heap or dump once, is collected, and pumped to the process plant to recover the gold and silver. As a result, the grade of the solution is relatively low, flow rates are relatively high, and the cost of the recovery system is higher than it needs to be. Improvements in these areas are becoming more accepted, as described later in the section, "Efficient Use of Leach Solution."

Scale Control

The chemistry for controlling the buildup of scale on pipelines and equipment has been known for a long time, but it took heap leach operators a long time to recognize that scaling problems must be controlled from the very first day of operation. Chemical companies have been good about servicing the industry and severe scaling problems are largely a thing of the past.

Detoxification of Heaps and Dumps

The cyanide solution that remains in permanent heaps must be removed according to criteria specified by the states when leaching operations cease. Removing the solution could be delayed many years until active leaching of the ore is completed. The heaps on reusable leach pads must be similarly cleaned, i.e., detoxified, before the leach residue can be removed from the leach pad. A summary of the recent state cyanide regulations has been compiled (Thatcher, et al., 1988; Smith and Struhsacker, 1988). ASTM's standard D 2036-82 defines standard analytical procedures for cyanides in waters.

Most cyanide species will decompose with time or they can be flushed out of a heap with cyanide free aqueous solution. Both of these processes take time. If the removal/destruction of cyanide must be speeded up, the addition of oxidizing chemicals is the most likely alternative. The chemical most used in the past for destroying residual cyanide has been calcium hypochlorite but it is gradually being replaced by hydrogen peroxide (Griffiths, 1988) or Inco's SO_2-air system (Devuyst, 1988). Other approaches to destroying cyanide have been developed. The simple use of ferrous sulfate is reportedly enough to minimize bird kills at solution ponds. At the opposite extreme, Homestake developed a bacterial system to meet their effluent requirements. The CNC process by Radian Corp. combines both chemical and biological unit operations to give clean effluents.

Actual detoxification experiences in Canada have been reported by Zaidi (1987).

HEAP AND DUMP LEACHING - THE FUTURE

The dump and heap leaching industry has matured but there is still room for improvement. Some of the improvements that are imminent or may occur in the future are described next.

Computer Simulation

Performing all the testwork that is required for designing a heap or dump leach operation takes much time and money. The testwork should be done, of course, but computer simulation of heap leaching will give management a preview of what can be expected and it can be used to help design the testwork. The simulation can be used to forecast the consequences of changes in ores or design parameters (Prosser, 1989).

Today's mineral processing models are fairly well developed and simulation packages have been described in the literature, but they tend to be academic and research-oriented; therefore, operators and engineering companies don't use them routinely. Popularizing the process models and simulation pack-

ages and making them user friendly will be necessary before they are widely used (Sastry, 1989).

Use of Oxygen

The use of oxygen to speed up leaching rates has reportedly become accepted practice in South African CIP and CIL circuits (Stephens, 1988). Kamyr, Inc. has been promoting its use in this country (Elmore, 1988). The leach rates are nearly proportional to the dissolved oxygen content of the leach solution. Although the use of oxygen under heaps or dumps has not been reported, it seems logical that it will be tried, especially in deep heaps and those that contain sulfides. Oxygen flow rates would be slow to allow maximum adsorption in the leach solution and to minimize losses to the atmosphere. The benefits may be increased recovery, passivation of preg robbing constituents, and faster leach rates.

The use of air rather than oxygen may offer cost advantages, i.e., power cost for blowers vs. cost of oxygen. And, if the heap has sulfides that generate heat and the resultant "chimney effect" within the heap, then air may be naturally aspirated through pipes at the base of the heap and the power cost for blowers may be reduced.

Air, however, will not raise the disssolved oxygen content of the leach solution as much as oxygen. Testing of the two systems needs to be done in column tests.

Leaching in Cold Weather

The prevailing thinking years ago was to heap and dump leach only during warmer weather so that solution spray systems did not freeze or cause a thick ice cap on the heaps. More recently, however, especially with the introduction of drip irrigation systems, year-round leaching in severe winter areas is gaining acceptance. Burying the drip irrigation lines in the surface of the heap prevents them from freezing and eliminates the build up of large amounts of ice on top of the heap. Deep heaps and dumps work better for this than shallow heaps because of the reservoir of heat they contain. Careful attention to engineering and construction details will make wintertime heap and dump leaching very acceptable.

If the ore contains sulfides, wintertime heap and dump leaching may be quite desirable because the heat generated within the pile will create an upward air flow or "chimney effect" which will naturally draw in fresh air through the sides of the pile. The chemical reactions are enhanced by the fresh source of oxygen and leaching rates will remain high. In fact, leaching rates in the winter could be faster than in the summer because of the chimney effect.

Crushing

Roller presses, recently introduced by KHD Humboldt Wedag Ag, produce a product that appears to have more microfractures than more crushed products (Esna-Ashari, 1988). The presses also appear to consume less energy than other crushers. For a heap leaching operation, the microfractures are advantages that will lead to higher recovery, faster leaching rates, and lower costs. Roller presses may also reduce the need for binder in agglomeration because the press forms flakes of aggregated ore that are fairly stable when exposed to a trickling cyanide solution. The gap between the rollers is typically 10 to 30 mm and the reduction ratio is about 3:1.

Better Binders

The binders that are used for agglomeration are alkalis, such as Portland cement. This makes it costly to use neutral or acidic leaching solutions because the pH is always rising. What is needed is a binder that will work in acidic media. Nalco has developed such a polymeric chemical and is testing it on a copper heap leach in Arizona. If cost effective, this should open the door to alternative lixiviants such as the halogen chemicals without ruining the permeability of the heaps and dumps.

Better Leach Pad Construction Materials

Geomembranes typically have slippery surfaces and so manufacturers have been building in a texture which increases the friction angle and therefore increases leach pad and heap stability. The geomembranes are relatively delicate, however, and puncturing them is easy to do. They also require field assembly of the relatively small shippable panels into the large monolithic sheets that must cover the entire leach pad. The field assembly is not always perfect and leaks that are difficult to locate can result. Methods or materials are needed to reduce the ease with which leaks can be made and the difficulty of repairing them.

One such material may be to use geocomposites such as Claymax. It is a sandwich of treated bentonite clay between polypropylene and polyester geotextiles. Field fabrication is not needed; the material self-heals if punctured; and reported permeability is lower than state regulations require.

Efficient Use of Leach Solution

Heap and dump leaching are slow processes that require large volumes of leach solution. The usual procedure is to percolate the solution through the heap once then pump it to a carbon or zinc recovery system to recover the gold and silver. This requires an expensive recovery system because large volumes of low grade solution are processed.

There is a gradual awakening to the fact that countercurrent heap leaching should be practiced just as countercurrent agitation leaching and countercurrent decantation are practiced. Passing the leach solution through the heap three times (low grade ore, middle grade ore, and fresh high grade ore) before pumping it to the recovery system will not only leach three times the tonnage with the same amount of solution but also give a higher grade preg solution.

Countercurrent heap leaching could also be com-

bined with intermittent spraying to achieve even greater use of the leach solution before it is pumped to the recovery system. Intermittent spraying means applying leach solutions to an area of the heap only 8 to 12 hr per day. During the remainder of the day, it is "resting" and the leach solution is being used to spray another area of the heap. Many column leach tests have shown that intermittent spraying does not usually reduce the rate of leaching compared to full-time spraying. If the rest period was many days or weeks, the rate of leaching would decrease. But the rate is not decreased if the rest period is only 12 to 16 hr. By using this concept, two to three times as much ore can be kept actively leaching with the same volume of leach solution that is required when leaching a single area full time. The primary advantage of this concept is that the size of the recovery system can be reduced and capital dollars can be saved.

Bacterial Heap Leaching

Bacterial oxidation of pyrite and other sulfides plays a very important role in biohydrometallurgy. It provides sulfuric acid and ferric ions which are needed in the heap and dump leaching processes as leaching agent and oxidant (Torma, 1989). Chemical lixiviants will not be replaced by bacterial leaching of heaps, but bacterial leaching will be used to supplement chemical leaching when acidic systems are used on sulfidic ores. This type of heap leaching will probably be used on the low grade portion of an ore body and the high grade ore will be treated by agitation leaching. Success will not come easily, but the interdisciplinary approach that operators have been forced to learn for heap leaching is a major step in the right direction (Holmes, 1988).

Alternative Lixiviants

Cyanide is the lixiviant of choice today. Thiourea has not been applied very much and thiosulfate even less so. However, companies are working on proprietary reagents containing bromine, or iodine, or chlorine. Halogen compounds have great promise but have not been widely accepted yet (Sergent, 1988). Perhaps that is because most of the ore deposits that have been developed so far are relatively clean and easy to treat. When companies are forced to consider deposits containing cyanicides, in spite of the high cyanide consumption, then alternative lixiviants will become more common. Pressures will also be brought to bear on operators to try them. The pressures will be imposed by the increasingly difficult chemical and physical nature of available ore deposits and by state regulators responding to public environmental demands.

Counteracting the use of alternative lixiviants will be the use of cyanide regeneration systems to keep cyanide costs under control. All of the systems use variations of a common theme: the cyanide solution is acidified; hydrogen cyanide gas is evolved; and the gas is reabsorbed for reuse. Not many of these systems are in commercial use today, but they have been proposed (Kitney, 1988; Witteck, 1986; Resource Technology, 1986). If the consumption of NaCN is >1.0 kg/ton of ore, the use of cyanide regeneration may be economic.

Another development that will maintain cyanide's position as the premier lixiviant for gold and silver is the low cost generation of cyanide systems at the mine site. Cyano Systems, Inc. claims that the direct production cost of cyanide from their modular system is about $0.53/kg. Amortization, maintenance, and fees will probably double that cost to ~$1.00/kg, which is still a very attractive price. The system is claimed to be safe and requires little operator or maintenance attention.

Use of Surfactants

The use of surfactants to increase recovery or leaching rate, or both, has had occasional successes. On the other hand, they have occasionally fouled carbon circuits. More definitive work needs to be done to determine which of the thousands of surfactants are helpful, and why. The prospect of an additional 5% extraction is the incentive.

Percolation Rate vs. Heap Height

A device is needed to determine the percolation rate through ores that are under pressure -- pressure that simulates heaps up to 200 ft deep. One of the most important economic questions every heap designer faces is, "how high can I build the heap and still get adequate percolation?". Column tests at 6, 10, 20, and 30 ft give useful information, but it would be risky to extrapolate this data to 100 or 200 ft. The "percolation under pressure" tests should not be performed quickly, i.e., within less than one day because the strength of agglomerates and even the mineralogical character of the ore itself changes with time within the heap leach environment (Kurtz, et al., 1987).

IN SITU LEACHING - PROGRESS TO DATE

The concept of in situ leaching is quite simple. In essence, two or more holes are drilled into an ore deposit. A leach fluid is injected into the injection holes and pregnant liquor is withdrawn from the production holes. The ore body is not disturbed.

In situ leaching enjoys certain advantages over more commonly accepted mining and milling methods. Primary among the economic advantages is the ability to recover values from deposits that are uneconomic for conventional mining methods due to depth, low tonnage, water saturation, ground support problems, or low grade. In addition, the capital and operating costs are generally lower; start-up times are faster; and the leaching operation usually has less impact on the environment. It is a way of processing small high grade or large low grade deposits that are too deep for open pitting but too small for underground mining. It is not generally applicable to surface deposits, but it could be used to extract extra values from deep dumps, pit walls, underground stopes, and subsidence zones (Hiskey, 1986).

In situ leaching has been used for the recovery of uranium, copper, salt, potash, and sulfur but not for gold and silver. In spite of the fact that re-

covering uranium and copper with acids mobilizes more ions than leaching gold ores with cyanide does, state regulators are more attuned to the fears of cyanide. Operators understand the resulting politics of cyanide and are hesitant to be the first to try cyanide in situ leaching for gold and silver.

However, cyanide is not the only lixiviant for in situ leaching gold and silver ores. For example, Jacobson and Murphy (1988) have presented a case for the in situ mining of argentite with a lixiviant of ammonium thiosulfate and ammonium carbonate. Halogen compounds also show promise for in situ leaching operations not only because they are less toxic than cyanide but also because they are effective oxidizing and complexing reagents for gold and silver. Jacobson and Murphy (1988) developed an iodine/iodide system for leaching deep gold placers. An electrowinning diaphragm cell regenerates the iodine as it recovers the gold and well field cleanup is accomplished by capturing residual iodine/iodide on a strong base anionic resin.

A common fear that in situ leaching operators must contend with is the escape of leach solutions, i.e., an excursion. If a leach fluid were injected into the deposit with no attempt to produce the pregnant liquor, then the fear of excursions and environmental damage would be great. But, if the volume of fluid recovered from production wells is slightly greater than the amount injected, which is always the case, there is a driving force imposed on the fluid in the ore zone and its direction of flow can be controlled. In situ operators have developed several lines of defense against fluid excursions beyond the boundaries of the ore zone. These include not only the driving force on fluids but also (1) guard wells into which water can be injected to drive leach solutions back to the ore zone, (2) monitor wells to warn of impending excursions, and (3) grout curtains around the perimeter of the ore zone to physically prevent excursions. Which of these defensive measures is needed is dictated by the stratigraphy, size, shape, depth, permeability, porosity, and mineralogy of the ore zone and by the chemistry of the system (Huff, 1988).

It is generally believed that in situ leaching must be done below the water table. This is desirable but not a necessity. In fact, most of the gold and silver ore bodies that have been mined are above the water table. Ore bodies above the water table can be operated within an artificial water table that is created by injecting water into barrier wells around the perimeter of the ore body. The use of polymers to form a grout curtain around the ore body is another technique for encapsulating the deposit so it can be leached within an artificial water table. Grout curtains have been used in the oil industry, but the spacing of polymer injection points has been <2 m. Tests need to be done at wider spacings, i.e., 8 m, to make grout curtains economically attractive for in situ leaching.

It is also believed that in situ leaching must be done in rubblized rock in order to have adequate permeability. The uranium sandstone deposits were an exception because their permeability was naturally quite high. But there is a growing belief, even for deposit permeabilities as low as 2 to 5 millidarcies, that rubblizing is not the best approach to take. Rather, the operator should use the same flow channels that Mother Nature used when forming the deposit. It is along these flow channels that the gold and silver mineralization was deposited and the most efficient use of lixiviants is to pump them only through these same channels (Wargo, 1988). Why break the rock and expose barren rock to the lixiviant? Keep the lixiviant only in the natural flow channels and the grade of the pregnant solution can be increased while minimizing the volume. Admittedly, this may require high injection pressures. The chemistry of the system must also be clearly understood and controlled to prevent the formation of unwanted precipitates that could plug the ore body. Overall, the advantages of leaching unrubblized rock will probably outweigh the disadvantages.

IN SITU LEACHING - THE FUTURE

The near-term outlook for in situ leaching of precious metals with cyanide is poor because of the perceptions held by the public and state regulators. It seems that they generally believe that anything injected into the ground is bad and that anyone who does so is guilty of pollution until proven innocent. It is the "proving of innocence" that deters most companies from using in situ leaching. They recognize that they will probably end up in court trying to prove their innocence and they fear the liability that may ensue.

On the other hand, in situ leaching with a non-cyanide lixiviant may be tried by some innovative company that has an ore body which is not economic with either open pit or underground mining. The economics of in situ leaching are very attractive on many ore bodies, even with two or three levels of redundancy to prevent excursions, and some company will try it for the money that can be earned. Success will be dependent upon a good working relationship with state regulators, a good ore body, and proper testing and implementation of all technical aspects of the project. The technical know-how is available today. The missing ingredient is a reputable company willing to give it a try.

CONCLUSION

Heap and dump leaching have reached the stage of mature technologies and they have done so within the past 10 years. They have been widely used because they are low cost technologies capable of processing low grade ores and because they can be put into production quickly. There have been many failures along the way because of insufficient attention was given to items such as heap building procedures, agglomeration of the ore, adequate testing, leach pad construction, geotechnical assessment of materials, and water balances. But the lessons have been learned, written about, and distributed to the industry through various media. Today the chances of success for a new venture are very high.

Those involved with heap and dump leaching are continuing to make improvements. The rate of leaching will be increased via the use of oxygen, opera-

tions in cold weather environments will be more common, new binders will be developed to allow the use of lixiviants other than cyanide in acidic systems, more efficient use of leach solution will allow reductions in capital cost, and computer modeling and simulation will allow a faster and more complete evaluation of new projects.

In situ leaching of precious metals has not been commercialized yet. The technical knowledge is available from operations in the uranium and copper industries, but the environmental concerns about using cyanide underground are pervasive. In situ leaching has the potential for being a very low cost technology, but it may sit on the shelf until lixiviants other than cyanide are accepted.

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